I. C. 6497 SEPTEMBER, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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MILLING METHODS AND COSTS AT THE MONTANA MINE CONCENTRATOR OF THE EAGLE-PICHER LEAD CO., RUBY, ARIZ.

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I.C.6497
September, 1931.

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MILLING METHODS AND COSTS AT THE MONTANA MINE CONCENTRATOR OF THE EAGLE-PICHER LEAD CO., RUBY, ARIZ.[1]

By D. E. Andrus[2]

INTRODUCTION

This paper, which describes the milling practice at the Montana mine concentrator of the Eagle-Picher Lead Co., is one of a series being prepared by the United States Bureau of Mines.

LOCATION

The Montana mine is situated on the eastern edge of the Oro Blanco mountains at an elevation of 4,300 feet. It is in the Oro Blanco mining district, Santa Cruz County, in the southern part of Arizona, about 4 miles north of the Mexican border, 30 miles northwest of Nogales and 80 miles southwest of Tucson. Amado, a station on the Southern Pacific Railroad, midway between Tucson and Nogales, and about 40 miles northeast of the mine, is the shipping point and is reached over an excellent highway.

HISTORY

The Montana group of claims was originally located in the 70's and was patented by L. Zeckendorf, in 1907, who retained the title until it was sold to the Eagle-Picher Lead Co.

In 1891, a 10-stamp mill, utilizing concentration and pan amalgamation methods, was built to treat the oxidized gold, silver, and lead ores. The mill was not successful and was abandoned in 1893.

In 1916 the construction of a second mill was started, but was not finished until the Goldfield Consolidated Co. acquired and completed it in 1917. This mill used flotation methods and operated during the remainder of 1917 and part of 1918. It produced 1,250,000 pounds of lead, 1,300,000 pounds of zinc, and 47,635 ounces of silver. During 1918 the Goldfield Consolidated Co., rather than meet a large payment due at that time and also owing to the scarcity and inefficiency of local labor, closed the mine and mill and gave up its option.

The property was idle until 1926, when an option was taken by the Eagle-Picher Lead Co., operating under the name of the Montana Mines Operations. A 250-ton capacity mill,

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[1] The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6497."

[2] Concentrator superintendent, Montana Mine Concentrator, and one of the consulting engineers, U. S. Bureau of Mines.

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using differential flotation methods was completed early in 1928 and has run intermittently since that time. The first operations of this plant lasted from May, 1928, until the water from the dams was depleted early in the same summer. After the summer rains made water available, the mill was started and operated until the latter part of January, 1929, when the mill was again shut down for lack of water. During this shut-down some changes were made, including the addition of conditioners ahead of the zinc circuit. Operations were again resumed after the summer rains in August, and in the mean time, a pipe line was constructed to obtain water from the Santa Cruz River, 15 miles distant, and thus insure an adequate water supply. The mill operated until the middle of April, 1930, when it was closed down owing to the low prices which prevailed in the metal market.

TRANSPORTATION

All concentrates are shipped to Amado by Coleman trucks, and the necessary supplies for the plant make up the load for the return trip. The trucks, which make a round trip in about 16 hours, are equipped with trailers, the capacity of one unit being 12 tons of lead concentrates or 11 tons of zinc concentrates. The hauling rate is $4.50 per ton or slightly more than $0.11 per ton-mile. This service has proved satisfactory.

WATER SUPPLY

Water is impounded by four dams, the storage capacity amounting to 40,000,000 gallons. During years of normal rainfall this amount is sufficient for mill operations, but for emergencies a pump and a pipe line were installed at Noria, on the Santa Cruz River, as previously stated. So far the installation has been used only one month as a test, as there has been sufficient water without it. The equipment consists of a 120-hp. Worthington 2-cycle Diesel engine, direct-connected to a Worthington duplex double-acting outside packed pump. The pipe line is 15 miles long and is constructed of 4-inch, heavy-duty, lap-welded pipe. During the month that the pumping plant was operated, the pump delivered water at the rate of 101.5 gallons per minute against a static head of 1,490 feet, a friction head of 585 feet, or a total head of 2,075 feet, at a cost of 25.8 cents per thousand gallons.

The mill pond is located back of the No. 1 dam and the amount of water stored in this pond is controlled closely by admitting water from the other fresh water dams as needed. This water flows by gravity from the upper dam, which is the reservoir into which the water from the Santa Cruz pipe line is delivered, but must be pumped to the No. 1 dam from the other two dams. Fifty gallons of fresh water per minute is an ample supply for all purposes.

POWER

Power is obtained by Diesel engines, one power plant serving both the mine and the mill. The plant contains six Fairbanks-Morse engines of the full Diesel type, and all engines are equipped with the same size heads, pistons, rings, and other interchangeable parts. They are operated by one engineer and one helper per shift and are under the direct supervision of the master mechanic. The mill requires 409 hp., which costs $0.0094615 per horsepower-hour. The sizes of the various engines and the service performed by each are as follows:

Size of engines and service performed by each

Engine number	Number	Rated Capacity, h.p.	Service Performed
1	3	180	Direct connected to a generator which furnishes 60-cycle 2-phase current at 220 volts for the mine hoist, mill pump, conditioners, and Wilfley pumps in the flotation circuit.
2	3	180	Connected by belt to a 2-stage Ingersoll-Rand compressor of 888 cubic feet capacity.
3	1	60	Connected by belt to a generator which is held in reserve for emergencies.
4	1		Direct-connected to a generator which furnishes power for camp lighting, refrigeration, and the motors which drive the pumps for the return of reclaimed tailings water.
5	2	120	Connected by belt to the line shaft which operates the crushing plant.
6	6	360	Connected by belt to the main line shaft of the mill.

ORE TREATED

The ore occurs in a vein which has a strike nearly east and west and a dip of 57° to the north. The ore body follows a conglomerate-diorite contact, and although it is usually contained in the conglomerate with the diorite forming the footwall it is also found within the diorite.

The primary ore is composed of mixed sulphide minerals and quartz gangue. The sulphide minerals comprise galena, sphalerite, and pyrite with subordinate amounts of chalcopyrite and tetrahedrite. The ore also contains some gold which is apparently free and not associated with the pyrite.

A microscopic examination of the ore was made by R. E. Head, United States Bureau of Mines, Intermountain Experiment Station, Salt Lake City, Utah, and his report to the company gives the following information:

> The microscopic examination of the Montana ore consisted in the study of briquetted mineral grains obtained by carefully panning the sample of ore submitted for the purpose. Several thousand individual particles were studied and the results obtained may therefore be considered as representing the mineralogical characteristics of the entire sample.
>
> The Montana ore contains a very small quantity of copper, which occurs chiefly in the form of chalcopyrite and is associated almost entirely with the sphalerite. Occasionally, inclusions of chalcopyrite will be found in the galena, but this form of association is relatively rare in the samples studied.
>
> The chalcopyrite occurs in the form of minute specks in the sphalerite and associated with it, in many instances, is tetrahedrite.

The tetrahedrite occurs in yet smaller specks than the chalcopyrite and in no instance were specks of the mineral seen that were larger than 300 mesh.

Tetrahedrite is the silver carrier in this ore and it is scarcely possible to expect liberation except by extremely fine grinding. Whether or not such a procedure is feasible is largely a question of economics and also depends on the relative flotability of the sphalerite and tetrahedrite in a slimed condition.

The ore at times contains considerable amounts of gouge-like colloidal material. This material has a marked tendency to float, and to carry zinc slimes with it into the flotation concentrates of the lead circuit.

PRESENT METHOD OF CONCENTRATING

The mill is built on a steeply sloping hillside. It is of galvanized iron and timber construction, with concrete retaining walls and concrete foundations for machines. Missouri concentrating practice, with the exception of jigs, was followed in the design of the flow sheet, and all machines with the exceptions of ball mill, Wilfley pumps and American filters are of the types developed in the Tri-State district.

Figure 1 gives the flow sheet of the crushing plant; Figure 2 presents the flow sheet of the concentrator.

Crushing

The ore. after passing through grizzlies, with 8-inch spaces, located underground, is hoisted in skips and dumped directly into the coarse-ore bin. From this bin it is fed, with water, to an 18-inch Blake-type crusher which delivers material of 2-inch maximum size. The crusher product is reduced to ⅜-inch maximum size by a set of 42-inch primary rolls which operate in closed circuit with a 4 by 8 foot trommel. In this circuit an 18-inch belt elevator returns the screen oversize to the rolls. A second closed circuit which comprises a set of 32 by 24 inch finishing rolls, an 18-inch elevator, and four 4 by 5 foot Leahy screens reduces the ore to 5-mm. size. The undersize products of the Leahy screens are deslimed in an 18-inch belt drag classifier. The overflow pulp is conveyed to a 50-foot Dorr thickener and the sands are delivered to the fine-ore bin.

The details of wearing parts of machines which operate in the crushing plant are given in the tabulation which follows.

Wearing part	Material	Life	Cost of wearing part, per ton of ore treated
Crusher jaws	-	31,156 tons of ore	-
Crusher cheek plates	-	9,132 tons of ore	-
Primary roll shells	Cast iron	24,801 tons of ore	$0.0242
Finishing roll shells	Forged steel	23,232 tons of ore	.0462
Trommel screens	-	9,934 tons of ore	.0044
Leahy screen cloth	0.105 diameter wire	14 days	.0051

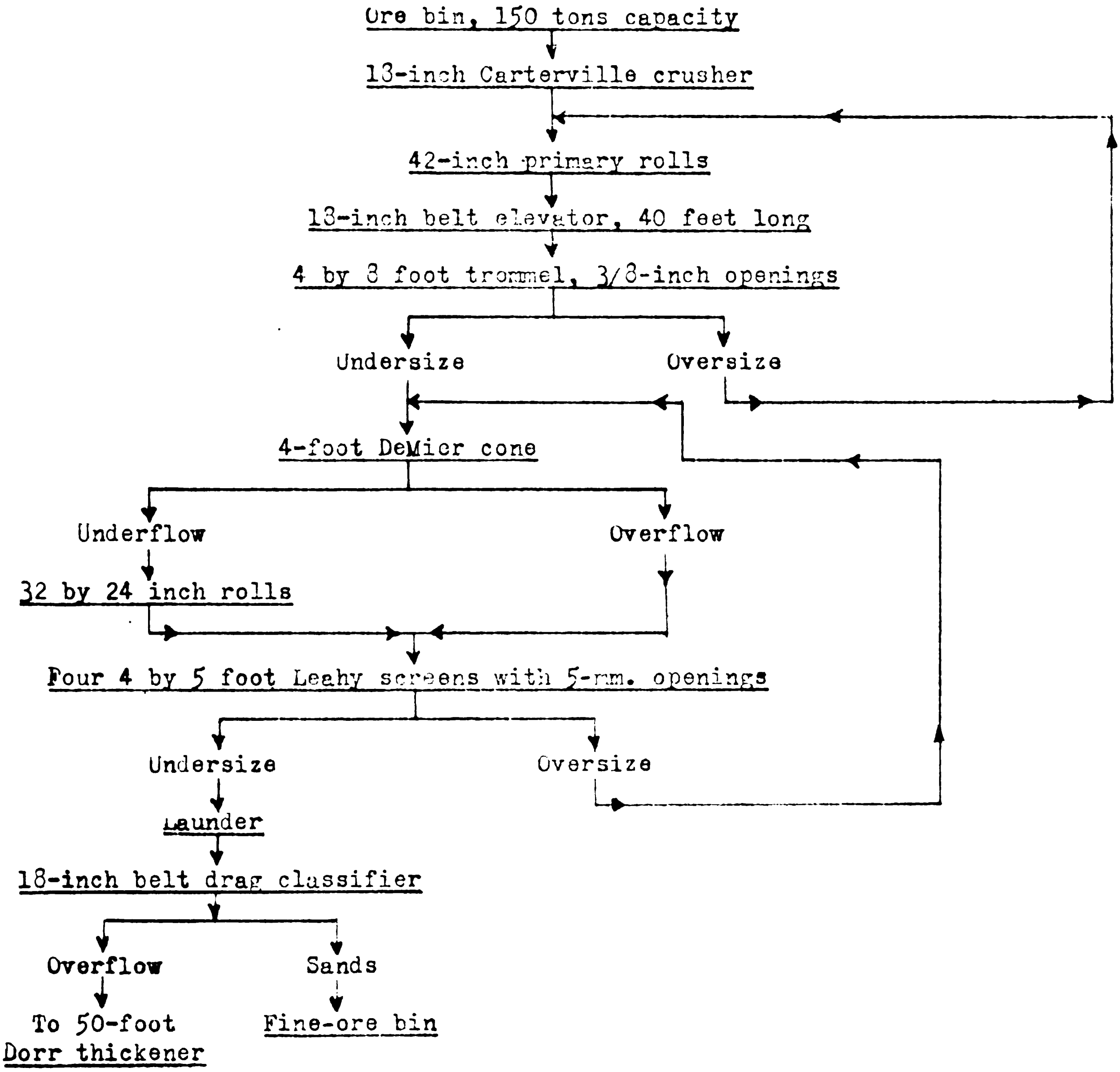
Ore bin, 150 tons capacity
13-inch Carterville crusher
42-inch primary rolls
18-inch belt elevator, 40 feet long
4 by 8 foot trommel, 3/8-inch openings
Undersize
Oversize
4-foot DeMier cone
Underflow
Overflow
32 by 24 inch rolls
Four 4 by 5 foot Leahy screens with 5-mm. openings
Undersize
Oversize
Launder
18-inch belt drag classifier
Overflow
Sands
To 50-foot
Dorr thickener
Fine-ore bin

Figure 1.--Flow sheet of the crushing plant

Classification and Fine Grinding

As considerable water is required to transport the crushed ore from the fine ore bin to the ball mill, it is necessary to dewater the feed pulp before it enters the ball mill, and a Dorr simplex classifier is used for this purpose. The classifier overflow pulp is delivered to the same 50-foot Dorr thickener that receives the overflow pulp of the desliming classifier of the crushing plant previously mentioned. The classifier sands are fed to an 8-foot by 36-inch Hardinge ball mill.

The ball mill is operated at a speed of 16 r.p.m. and with a ball load of 16 tons. Balls are cast in Tucson and replacements are made by the addition of 3-inch sizes added to the ball mill through the feed scoop during the day shift. The balls cost $44 per ton delivered at the mill; ball consumption amounts to 3.19 pounds per ton of ore ground under present conditions. The consumption and the cost of ball-mill liners are not known, as but one set of breast liners has been replaced to date. This replacement was made after the mill had ground 70,000 tons of ore.

The Hardinge mill operates in closed circuit with a belt drag classifier the belt of which travels at a speed of 56 feet per minute. This classifier has insufficient capacity and operates somewhat similar to a drag conveyor when loaded. The normal overflow solids contain approximately 30 per cent of minus 200-mesh material and considerable material that is plus 48-mesh.

Flotation

The thickened pulp from the 50-foot Dorr thickener joins the classifier overflow of the ball-mill circuit and both are conditioned in a 6-foot Denver machine. Sodium carbonate is added to the ball-mill feed pulp, by means of a dry feeder, in sufficient amount to produce a pH value between 8.0 and 8.3 when the pulp reaches the head of the lead circuit. Zinc sulphate, sodium sulphite, and thiocarbanalid reagents are also added to the ball-mill feed pulp launder, and just below the point where these additions are made, sodium cyanide, and sodium silicate are added each in the form of a 10-per cent solution. Ethyl xanthate and cresylic acid are added to the launder which conveys the middlings of the lead circuit to the feed elevator which in turn feeds the conditioner at the head of the lead circuit.

Sodium carbonate, added mainly to produce the desired alkalinity, also gives a selective action to some degree in the flotation circuit. Zinc sulphate, sodium cyanide, and sodium sulphite act as depressors of sphalerite in the lead circuit. The relative amounts of each are changed as needed to meet varying conditions in the character of the ore treated. The function of the sodium silicate is to disperse the fine gangue material of the gouge and thus prevent its tendency to float, and to carry zinc fines mechanically, into the lead concentrates.

After the pulp is conditioned it is treated in two primary Butchart flotation machines which operate in parallel, each treating one half the total flotation feed. These machines produce finished lead concentrates and tailings products, which are again treated in two secondary Butchart flotation machines, with additional xanthate and cresylic acid reagents. The secondary machines operate as scavenger cells and produce lead middlings, as froth products, which are returned to the conditioner at the head of the lead circuit. The tailings are conveyed to two 6-foot conditioner tanks at the head of the zinc circuit by a 3-inch Wilfley pump.

The reagents used in the zinc circuit are added to the sump which feeds the Wilfley pump. Lime is added by means of a dry feeder in sufficient quantity to produce a pH value of 8.8, the usual amount required being about 1.0 pound per ton of ore. Copper sulphate, 0.64 pound per ton of ore, is fed by a stream of water directed upon copper sulphate crystals which are held ina copper container. The saturated copper sulphate solution leaves at the bottom of the container. Sodium aerofloat, 0.025 pound per ton of ore treated, and sufficient cresylic acid to produce the froth desired, are also added.

After the pulp is conditioned it is distributed to two primary Butchart flotation machines which operate in parallel. These machines produce final zinc concentrates and tailings products. Additional amounts of ethyl xanthate and cresylic acid are added to the tailing pulps, after which they are again treated in two secondary Butchart flotation machines operated in parallel. The secondary machines operate as scavenger cells and produce zinc middlings, as froth products, which are returned to the conditioning tanks at the head of the zinc circuit by a 3-inch Wilfley pump. The tailings of the secondary machines are final waste products and are conveyed to the tailings dam by a 4-inch Wilfley pump.

As a portion of the silver content of the ore is intimately associated with the sphalerite, the lead circuit is operated to float about 10 per cent of the total zinc with the lead concentrates. The rejection of this zinc and the production of cleaner lead concentrates is accompanied by a marked decrease of the silver and copper contained in the lead concentrates and a corresponding increase of these metals in the zinc concentrates. The figures which follow illustrate the effect of floating zinc with the lead on the silver and copper contents of the lead concentrates.

	Lead, per cent	Zinc, per cent	Copper, per cent	Silver, ounces per ton
Lead concentrates	72.0	4.0	1.0	22
Lead concentrates	56.9	9.6	2.3	62

The primary flotation machines are operated with the gates, which control the pulp levels, down and the secondary machines are operated with the gates raised about five notches.

The poor classification obtained in the grinding circuit, as previously noted, causes the sending of considerable amounts of coarse material to the flotation circuits. This condition results in excessive wear on the impellers of the flotation machines and also causes choking in the conditioner tank at the head of the lead circuit, which makes necessary the draining of its contents to the feed box of the ball mill once every week. Flotation-machine impellers are renewed on the primary and secondary cells after handling 10,000 and 7,100 tons of ore, respectively. The cost of impeller renewals amounts to $0.005 and $0.007 per ton of ore treated in primary and secondary cells, respectively.

Runners, die rings, and follower plates of the Wilfley pump, which conveys lead circuit tailings, have a life of 3,770 tons of ore handled, whereas the case averages 13,748 tons before renewal. The cost of renewal parts for this pump amounts to $0.0107 per ton of ore milled. The cost of renewal parts for the Wilfley pump which returns the zinc middlings froth to the head of the zinc circuit amounts tl $0.0035 per ton of ore milled.

The coarse material contained in the flotation feed pulp does not, fortunately, prevent satisfactory recoveries being made in the flotation circuits. This condition is probably due to classification and selective grinding of the sulphide minerals.

DISPOSAL OF TAILINGS

The tailings from the zinc scavenger cells are conveyed to a sump and from there pumped to either of two tailings dams located 800 feet distant and directly below No. 1 fresh-water dam. The pipe line used for this purpose is served by a 4-inch Wilfley pump and is made of 4-inch, flange-connected, heavy-duty pipe. The flange connections permit the joints to be readily disconnected in case of choking or installing new lengths of pipe. The sides of the dam are maintained by two men during the day shift. The launder discharge points are changed as required on the two other shifts by the pump men, in addition to their usual pump duties, which include the return of reclaimed water from the tailings pond. The reclaimed water from the tailings pond amounts to 75 per cent of that contained in the tailings pulps; it is returned to No. 1 fresh-water dam by either of two 3-inch, motor-driven American centrifugal pumps. The case of the 4-inch pump which serves the tailings pipe line handles 15,219 tons of solids before renewal; the follower plates, runners, and die rings average 8,503 tons of solids before replacing. The cost of pump replacements is $0.00918 per ton of solids handled.

Filtering

The finished concentrates pulps from the lead and zinc flotation circuits are conveyed directly to their respective 6-foot, 4-disk American filters by 3-inch Krogh pumps. The pulps enter the filter reservoirs through the bottom, as this method of introducing the pulps tends to maintain suspension of the solids and thus increases the efficiency of filtering. Vacuum is normally maintained around 20 inches of mercury by an 18 by 6 inch Ingersoll Rand pump which operates at a speed of 274 r.p.m. Palma twill filter bags are used, which cost $0.67 each delivered to the mill and which have an average life of 29 days of service.

SAMPLING

All samples are taken by hand at 30-minute intervals. The concentrates are delivered by the filters to 14-foot concentrates cars which in turn deliver them to concentrates bins. The concentrates samples are taken from these cars by means of pipe samplers. Six pipe samples are removed from each carload of concentrates. Half of the total sample is placed in an air tight container to be used for the moisture determination and the other half is reserved for the usual chemical determinations. Weights of concentrates produced per shift are calculated from the number of carloads delivered to the bins and by calibration of the bins.

The weight of mill heads is calculated from the number of skip loads hoisted from the mine and from calibrations of the coarse and fine ore bins. All concentrator samples are composited for each shift, and the shift samples are assayed for gold, silver, lead, zinc, and copper.

The mill assays and weights of concentrates have closely checked those obtained by the smelters. As a rule, the mill weights are a fraction of 1 per cent less than the smelter

CONCENTRATOR DATA

Table 1 presents metallurgical data for various periods during 1929 and 1930. The marked improvement shown in March, 1930, as compared to October, 1929, is due to changes in flotation reagents and to longer experience of the operators with the flotation machines used.

Table 2 gives screen-assay analyses of final concentrates products and Table 3 presents a screen-assay analysis of the final tailings.

Table 4 gives estimated power requirements for the various machines operated in the plant.

Table 5 gives a summary of milling costs for the period October 1, 1929, to April 1, 1930. Table 6 shows unit costs for March, 1930, and Table 7 gives the distribution of power used for the period October 1, 1929 to April 1, 1930.

Tagle 1.- Metallurgical data for various periods during 1929 and 1930

	October, 1929	March, 1930	October 1, 1929 to April 1, 1930
Total ore milled, tons	7,049	6,149	41,246
Operating time, per cent	89.50	94.00	91.12
Ore treated per 24 hours of operating time, tons	254	252	258
Analysis of heads:			
Gold, ounces per ton	0.060	0.077	0.071
Silver, ounces per ton	8.57	8.02	8.17
Lead, per cent	5.94	5.89	5.13
Zinc, per cent	7.69	6.47	5.98
Copper, per cent	.452	.335	.390
Total lead concentrates produced, dry tons	647.20	597.72	3,435.06
Lead concentrates produced per 24 hours operating time, dry tons	23.32	24.46	21.47
Moisture content of lead concentrates, per cent	7.90	8.00	7.67
Ratio of concentration in lead circuit, tons into 1	10.89	10.89	12.01
Indicated recoveries in lead concentrates, per cent:			
Gold	65.80	82.19	78.31
Silver	67.70	78.61	75.49
Lead	88.40	95.79	92.59
Zinc	13.05	15.97	14.02
Copper	56.40	66.13	65.33
Analyses of lead concentrates:			
Gold, ounces per ton	.438	.654	.674
Silver, ounces per ton	63.28	64.88	74.02
Lead, per cent	57.29	58.11	57.05
Zinc, per cent	10.93	10.64	10.06
Copper, per cent	2.781	2.28	3.094
Total zinc concentrates produced, dry tons	747.42	545.95	3,215.15
Zinc concentrates produced per 24 hours of operating time, dry tons	26.93	22.34	20.09
Moisture content of zinc concentrates, per cent	10.30	9.00	9.58
Ratio of concentration in zinc circuit based on original heads, tons into 1	9.43	11.28	12.83
Indicated recoveries in zinc concentrates, per cent:			
Gold	14.25	8.10	7.89
Silver	18.10	10.91	11.92
Lead	6.80	2.68	4.23
Zinc	69.50	70.58	68.29
Copper	23.70	13.40	14.41
Analysis of zinc concentrates:			
Gold, ounces per ton	.082	.070	.07
Silver, ounces per ton	14.64	9.86	12.40

Tagle 1.- Metallurgical data for various periods during 1929 and 1930 - Continued

	October, 1929	March, 1930	October 1, 1929 to April 1, 1930
Analysis of zinc concentrates - (Cont'd):			
Lead, per cent	3.79	1.78	2.79
Zinc, per cent	50.46	51.45	52.36
Copper, per cent	1.013	.506	.720
Indicated tailings losses, per cent:			
Gold	20.00	9.68	13.78
Silver	14.10	10.46	12.58
Lead	4.15	1.53	3.16
Zinc	17.60	13.45	17.68
Copper	19.95	20.48	20.24
Analysis of tailings:			
Gold, ounces per ton	.014	.009	.011
Silver, ounces per ton	1.49	1.03	1.22
Lead, per cent	.35	.11	.19
Zinc, per cent	1.87	1.07	1.26
Copper, per cent	.100	.084	.095
Alkalinity in lead flotation circuit, pH value	-	8.2	8.0
Alkalinity in zinc flotation circuit, pH value	-	8.8	8.8
Consumption of balls per ton of ore, pounds	2.70	2.17	3.19
Solids in classifier overflow pulp, per cent	28.2	31.0	29.3
Reagents used per ton of ore treated, pounds:			
Lime	1.200	1.200	.995
Sodium carbonate	.385	1.28	.735
Ethyl xanthate	.050	.050	.057
Thiocarbanalid	None	.095	.028
Aerofloat	.035	.026	.031
Cresylic acid	.217	.190	.150
Pine oil	.030	.028	.042
Sodium cyanide	.544	.216	.219
Zinc sulphate	1.000	.850	.640
Sodium sulphate	None	.095	.028
Copper sulphate	.65	.65	.641
Sodium silicate	.48	.48	.258

Table 2.- Screen-assay analyses of final concentrates products

Screen size, mesh	Weight, per cent		Assays					
			Ounces per ton		Per cent			
	Individual	Cumulative	Gold	Silver	Lead	Zinc	Copper	Insoluble
Lead concentrates:								
Plus 40	1.50	1.50	0.90	31.0	4.3	4.9	0.60	40.8
Minus 40 plus 60	1.50	3.00	.90	42.0	40.1	10.2	2.67	15.6
Minus 60 plus 100	4.00	7.00	.88	113.6	52.1	10.7	5.27	9.7
Minus 100 plus 150	10.00	17.00	.80	99.5	48.5	10.6	2.19	6.3
Minus 150 plus 200	12.00	29.00	.75	76.0	54.7	10.5	3.11	2.6
Minus 200	71.00	-	.63	52.6	60.2	9.3	2.10	1.4
Composite	100.00	-	.679	62.05	56.90	9.57	2.34	3.17
Zinc concentrates:								
Plus 40	0.30	0.30	.02	5.20	4.20	10.50	.32	-
Minus 40 plus 60	1.40	1.70	.16	23.52	1.60	49.50	2.10	-
Minus 60 plus 100	7.20	8.90	.15	20.50	1.40	55.98	1.80	-
Minus 100 plus 150	17.10	26.00	.11	10.60	1.05	54.62	.70	-
Minus 150 plus 200	14.00	40.00	.07	9.20	0.80	53.40	.80	-
Minus 200	60.00	-	.08	8.60	2.45	52.50	.42	-
Composite	100.00	-	.089	10.08	1.89	52.77	.64	-

Tagle 3.- Screen-assay analyses of final tailings

Screen size, mesh	Weight, per cent	Analyses per cent		Per cent of total metals	
		Lead	Zinc	Lead	Zinc
Plus 40	2.60	1.35	1.17	35.05	2.50
Minus 40 plus 60	16.70	0.10	0.80	16.61	11.50
Minus 60 plus 100	20.60	.01	0.11	2.68	1.87
Minus 100 plus 150	15.50	.01	0.04	1.54	0.51
Minus 150 plus 200	8.60	.01	0.21	0.86	1.49
Minus 200	36.00	.02	2.77	43.30	82.58
Composite	100.00	.099	1.207	100.04	100.45

Table 4.- Estimated power requirements of the various machines with friction losses included

No.	Machine	Estimated horsepower	
	Crushing plant:		
1	Carterville crusher, 18-inch	20	
1	Primary rolls, 42-inch	15	
1	Coarse elevator, 18-inch	8	
1	Trommel, 4 by 8 foot	4	
4	Leahy screens, 4 by 5 foot	4	
1	Fine elevator, 18-inch	8	
1	Hercules rolls, 32 by 24 inch	15	
1	Belt deslimer	7	
1	American centrifugal pump, 3-inch	6	
	Total for crushing plant		87
	Fine grinding:		
1	Hardinge ball mill, 8-foot by 36-inch	135	
1	Dorr simplex dewatering classifier	3	
1	Belt drag classifier, 18-inch	5	
	Total for fine-grinding circuit		143
	Flotation:		
1	Feed elevator, 12-inch buckets	5	
1	Lead circuit conditioner, 6-foot	3	
8	Butchart flotation machines	64	
1	Wilfley sand pump for lead circuit tailings, 3-inch	10	
2	Zinc circuit conditioners, 6-foot	6	
1	Wilfley sand pump for return of zinc middlings, 3-inch	8	
1	Krogh pump for lead concentrates, 3-inch	5	
1	Krogh pump for zinc concentrates, 3-inch	6	
1	Dorr thickener, 50-foot	5	
1	Butchart diaphram pump, 4-inch	2	
	Total for flotation		114
	Filters:		
2	American filters, 6-foot 4-disk, including vacuum and filtrate pumps	25	25
	Water supply:		
1	Gould centrifugal pump, 6-inch	18	
1	American centrifugal pump for return of water from tailings dam	10	
	Total for water supply		28
	Disposal of tailings:		
1	Wilfley sand pump, 4-inch	12	12
Grand total			409

Tagle 5.- Summary of concentrator costs for period October 1, 1929 to April 1, 1930

Ore milled, tons	41,246
Lead concentrates produced, tons	3,435
Zinc concentrates produced, tons	3,215

	Labor	power[2]	Supplies, renewals, and repairs	Total	Per cent of total cost
Crushing and screening	$0.1493	$0.0765	$0.0954	$0.3212	14.59
Grinding and classifying	.0233	.1258	.1029	.2520	11.45
Flotation	.1347	.1003	.3631	.5981	27.16
Filters	.0204	.0219	.0080	.0503	2.28
Disposal of tailings	.0408	.0105	.0110	.0623	2.82
Water supply, new and return water	.0506	.0241	.0008	.0755	3.42
Total direct cost	$0.4191	$0.3591	$0.5812	$1.3594	
Miscellaneous[1]	.1675	-	.6751	.8426	38.27
Total cost	$0.5866	$0.3591	$1.2563	$2.2020	100.00

[1] Includes assaying, operation of pumping plant and pipe line from river when used, shop, and mill share of the general overhead expense.

[2] Power costs $0.0094615 per horsepower-hour.

Table 6.- Summary of unit concentrator costs for March, 1930

Ore milled, tons 6,149

	Total cost	Cost per ton of ore milled	Per cent of total cost
Labor[1]	$2,931.62	$0.4765	33.59
Power	2,448.55	.3982	28.24
Balls	293.70	.0477	3.38
Reagents	1,960.52	.3188	22.61
Wearing parts estimated[2]	1,094.52	.1781	12.18
Total	$8,728.91	$1.4193	100.00

[1]

Average number of men per shift	7.22
Tons of ore milled per man-shift	9.89
Tons of lead concentrates produced per man-shift	0.8239
Tons of zinc concentrates produced per man-shift	0.7712

[2] Estimate of cost of wearing parts per ton of ore milled follows:

Crusher	$0.0050
Primary rolls	.0242
Finishing rolls	.0462
Trommel	.0044
Screens	.0050
Liners	.0200
Wilfley pumps	.0233
Impellers	.0120
Filter bags	.0080
Other iron (including cast launders, scoop lips, elevator buckets, etc.)	.0300
Total	$0.1781

Table 7.- Distribution of power for period October 1, 1929 to April 1, 1930

	Horsepower-hours per dry ton milled	Per cent of total power
Crushing and screening	8.11	21.33
Ball mill and classifiers	13.35	34.94
Flotation	10.63	27.96
Filtering	2.32	6.10
Water supply	2.55	6.71
Tailing disposal	1.12	2.96
Total	38.08	100.00

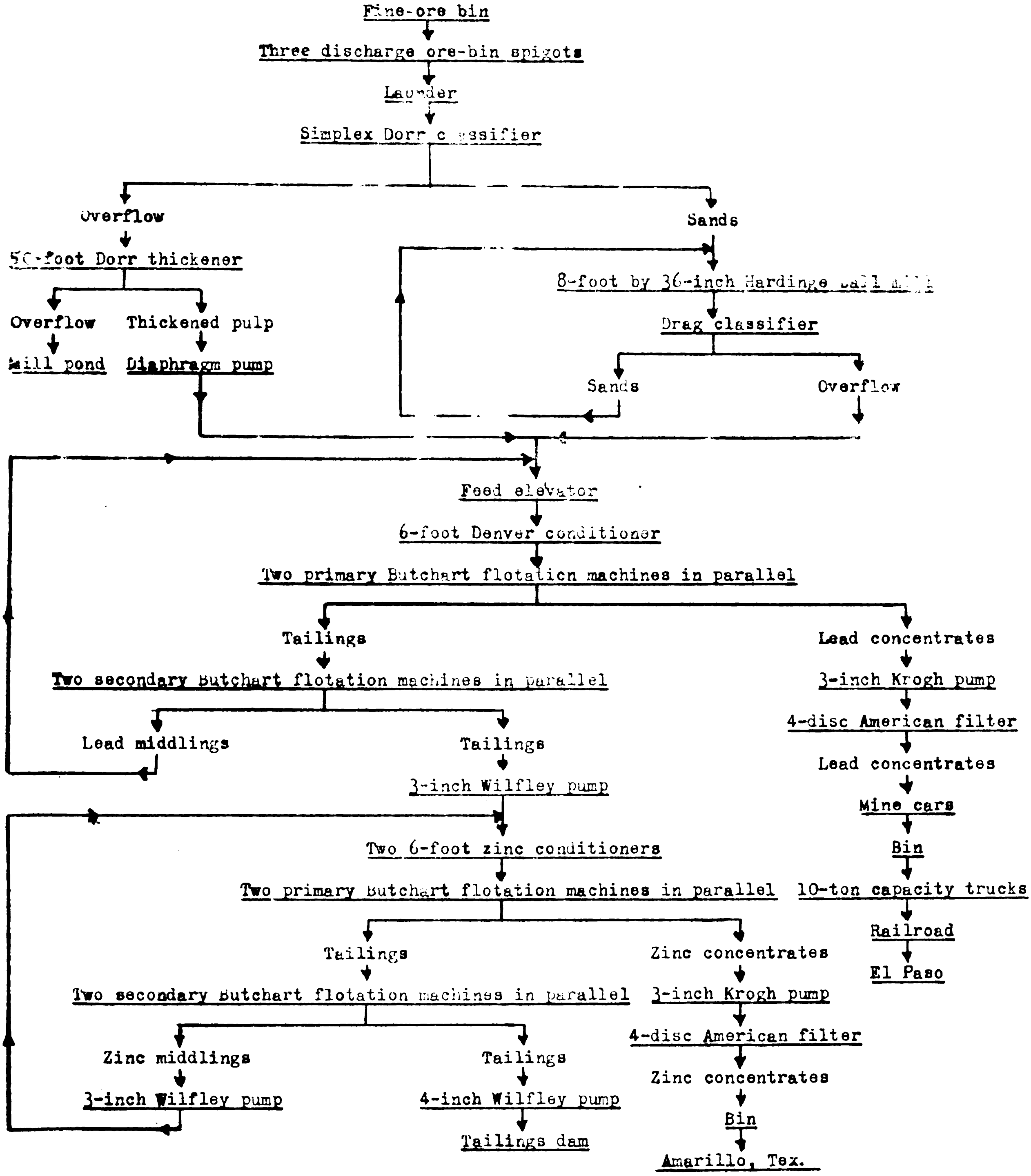

Figure 2.--Flow sheet of the concentrator